

By Dr.-Ing. C. Eymann of Walsum

CPYRGHT

In the last few years the "iron-coke" process has been developed by the Thyssen Gas- und Wasserwerke G.m.b.H. and the Bergwerksgesellschaft Walsum G.m.b.H. and up to the present time more than 12,000 tons of iron-coke has been prepared mainly at the Prosper coke ovens in Bottrop. This iron-coke has been used partly in a slagging gas producer at the Thyssen works, it has been intermittently tested in a large blast furnace at the August-Thyssen iron works and it has been finally given a continuous trial over a prolonged period and as a 100% charge at the Friedrichshutte A.G. works at Herdorf (Sieg). Recent developments have shown that a highly bituminous coal, i.e. a gas coal which cannot be made into a satisfactory blast furnace coke by itself, can be used successfully as a binder when mixed with 40% of finely ground iron ore and suitably coked. When mixed with a suitable iron ore this coal can be used to produce a solid large-sized iron-coke with an Fe content after coking of 35%. At the same time, due to the reaction between the oxygen of the iron ore and the coal a greater quantity of gas is produced which entails the consumption of coking coal during the coking operation.

The recovery of valuable by-products, such as tar and benzol is no lower when a fine ore of difficult reducibility is used than the production of iron-coke, the recovery of ammonia is less. The content of hydrocyanic acid in the gas produced during the distillation process is markedly less.

The recovery of valuable by-products, such as tar and benzol is no lower when coke and the consumption of fuel (gas) during this operation are later offset in the smelting of the iron-coke in the blast furnace when both ore and coke are saved. The equivalent recovery of ordinary coke in the iron-coke smelting process for the production of pig iron is better than in the normal operation of coking the usual grades of coal and then using this coke in the usual manner. This is because although there is a loss of efficiency in the coking chamber when iron-coke is produced, the intimate contact between ore and carbon in the smelting operation increases the metallurgical efficiency here. The larger proportion of iron-coke can be charged in the form of pieces of over 15 mm size, the remainder can be used as coke breeze in the sintering process. Here also the heat of reduction of the iron can to a greater or less extent replace the heat of combustion. The larger size of the fine ore in the coke breeze is advantageous here, and in order to take full advantage of the coke-equivalent in iron-coke care must be taken to see that no oxidation of the reduced iron, i.e. no roasting of the iron-coke takes place during sintering.

In the mixing of fine coal and fine ore the ore to some extent forces its way into the spaces in the agitated fine coal, thus the weight per unit volume of coking coal in the coal-ore mixture is increased due to the entrainment of the heavier ore. The quality of the coke is thus improved in a manner similar to stamping or compressing. In the production of iron-coke therefore the reduction in the coal capacity of the coke oven does not correspond exactly with the ore content in the mixture.

If a coarser size of fine coal be employed it is possible to produce a very hard grade of iron-coke. This, however, can lose its strength at a higher temperature. Such an iron-coke contains clusters of ore (Fig. 1), which on being heated to the higher temperatures ruling in the blast furnace cause the localised production of concentrates of oxygen which destroy the coke skeleton. Local concentrations of ore weaken the structure of the coke and favour the collapse of the coke particles, and iron-coke of this sort can undergo collapse during its passage down the blast furnace shaft. The preparation of iron-bearing coke has been undertaken experimentally in several places in the last ten years using medium bituminous coal. In cases where the work has not been fully successful the difficulty has usually been traced to the formation of "nests" of ore in the mixture. The strength of iron-coke as determined at room temperature gives no indication of its behaviour in the blast furnace, for good iron-coke must maintain its strength as it passes down the furnace stack. This can only be obtained by the proper fine milling of the coal, the use of finely-milled ore and uniformity of mixing the two (Fig. 2).

The mixing of fine ore or flue dust with finely-ground coal is facilitated by the addition of a little oil or something similar. At the same time the production of gas is enhanced and also the weight of the mixture which can be poured into a coke oven and settled therein satisfactorily is increased. The addition of oil produces a stable ore-coal mixture in which the two constituents remain uniformly mixed in spite of repeated unloading, re-loading and general agitation. Therefore, there is no tendency to form local concentrations of ore in the mixture, even with a high percentage of ore present, and the effects which this can cause are therefore avoided.

In an iron-coke made with fine-ground coal and fine ore there are no ore "nests". Oxidation can only take place on the surface for in the interior of each coke particle metallic iron is combined and therefore protected against further oxidation, i.e. against a reduction in the coke-equivalent (Fig. 3). If these conditions are not fulfilled however then small concentrations of oxidation can be set up when the iron-coke is in storage and these "nests" of oxidation can have the same effect as ore "nests", when the iron-coke passes down the blast furnace shaft the coke skeleton is disrupted.

A condition for the production of good iron-coke is the use of a coal with the right bitumen content. Coals which are low in bitumen content can only be combined with a little fine ore, conversely highly bituminous coals can carry a greater quantity of ore. In the latter case the strength of the coke is increased so that a smeltable iron-coke can be made from a mixture of finely-ground coal and a suitable ore without the addition of a "fat" coal. To make iron-coke in this way has been the main object of our work.

For the production of iron-coke we utilised the Walsum coal which contains between 33 and 34% volatiles (dry analysis) and a magnetite dust. The coke prepared from this coal is normally of a small size and a brittle character. By the addition of 5 - 7% by weight of coke breeze, which is 10 - 14% by volume the coke made from Walsum coal becomes larger-sized and its Abrasion Index rises by about 7 points. If in place of the coke breeze the same volume of magnetite dust is added, i.e. 20 - 30% of magnetite by weight then the coke loses its fragility, it becomes coarse in lump size and in strength and structure it resembles blast furnace coke

(Fig. 4a and 4b). By the addition of the iron ore the thermal conductivity of the compressed ore-coal mixture is enhanced. This is one of the reasons why a hard iron-coke can be made from a highly volatile coal (Fig. 5). The impregnation of coal with ore is responsible both for the increased bulk density of the ore-coal mixture and for the improved hardness and strength of the resulting coke.

The type of iron ore used plays a decisive part in the production of iron-coke. Easily-reduced ores quickly lose their combined oxygen during coking and thus neutralise the bitumen content of the coal. These ores thus lower the recovery of the by-products, the yields of tar and benzole are both reduced. Ores which are difficult to reduce on the other hand have little influence upon by-product recovery, they simply act as inert substances during coking. They can only be added to coal in appreciable quantities if the coal used is sufficiently bituminous.

Fig. 6 shows the relationship between the gas yield and the length of coking time during the coking of Walsum coal with and without the addition of iron ore. The reduction of the ore, as the gas yield indicates, sets in early and in the positions in which the coke quickly reaches a high temperature, for example in the layers of coke next to the kiln walls. The figures on the curves give the calorific value of the gases generated in kilocalories per kilogram of coal. When Walsum coal is coked by itself from the 18th hour onwards the production of gas falls sharply, whereas when iron ore is included in the charge the production of gas proceeds at the same rate as at the beginning and is not complete until after 20 hours of coking. A few tests were carried out in which coking was continued for a further 4 hours to examine the further gas yield, the gas yield continued practically undiminished in these cases. In the production of ordinary coke the reaction was practically complete after 20 hours. The calorific value and volumes of gas produced up to this point have been compared with the same values for the production of gas during the preparation of iron-coke. The difference in these figures can be ascribed to the reduction gas. The actual volume and analysis of this reduction gas cannot be accurately assessed as it reacts with the other gases generated, which reaction is considerably influenced by the catalytic action of the iron. The reduction gas which is generated in the 20th to 24th hours of coking iron-coke also contains the gas from the final after-gassing of coke as ordinarily produced. In general however ordinary coke is not over-coked in this manner. In the presence of iron ore however the release of gas and with it the reduction of the ore proceeds so vigorously that an extension of the normal coking time is not necessarily uneconomic. The loss of combustion heat in the release of reduction gas into the coke oven gas is offset by the enhancement of the reduction reaction. During the extra coking time the heat input to the coke oven amounted to 100 kilocalories per kg of dry coal. The production of the reduction gas gave a credit of 145 kilocalories per kgm of coal, i.e. during this time 45 kilocalories per kg represent the heat-giving value of the reduction gas.

If the heat balance of the coking operation is drawn up it is apparent that the firing of the oven is dependent upon the time of coking, i.e. on the degree of ore reduction. The sensible heat of the reduction-coke, which is lost when the oven is pushed, must be regarded as heat input. The sensible heat and calorific value of the coked material is higher than calorific value and sensible heat before the reaction has taken place. In this case a heat displacement has taken place from the coal side to the ore side. There are various reasons which can be advanced to account for

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this. For example if the fuel consumption for firing the oven be determined for the production of ordinary coke and then under the same conditions for the production of iron coke the displacement can be calculated by difference. However, the specific fuel consumption for the coking of coal is not a fixed quantity but is dependent upon the way in which the process is carried out. If a lower specific firing rate be used then the heat displacement drops from 585 kilocalories down to 575 kilocalories if the coking time be short and down to 540 kilocalories if the coking time be longer. Under these conditions, therefore the fuel consumption for the production of iron coke increases. Results of this kind are not unusual. In individual tests specific fuel consumptions of 540 kilocal/ton of coal have been recorded, and even lower values than this. In ordinary practice results of this kind can be obtained only if a continuation of the firing is avoided and the coking operation brought to an end the moment the reaction is complete. If the coke is heated for longer than this then one part of the unnecessarily applied heat is lost in the chamber walls and in the regenerator while the other part raises the final temperature of the coke unnecessarily high and is lost by radiation and in the chimney gases when the oven is pushed and the coke quenched. In practice it is generally impossible to bring the cycle time so well into line with the actual coking time. The coal is slightly damp and the charged weight varies within certain limits so a certain degree of overheating of the coke cannot generally be avoided. In addition the temperature of the oven walls is not fully uniform, therefore to make certain that all the charge is coked some of the coke has to be overheated and it is well-known that for these reasons a not inconsiderable proportion of the applied heat has to be wasted.

With the coupling together of the reactions of coking the coal and reducing the iron ore however it is possible to bring about a considerable improvement in the efficiency of heat utilisation. Any heat applied to the ore-coal charge after the coking reaction and the formation of a strong iron-coke will in part, as shown above, be utilised in the formation of gas and the reduction of iron oxide. The reduction of the ore acts as a brake on the temperature as this reaction starts when the temperature is between 700 and 800° C. In the production of iron-coke therefore an improved heat utilisation is possible and a reduction in external fuel consumption, and this explains the improved thermal conditions when coal is coked in presence of magnetite ore. Unlike the normal coking procedure, a standover in the oven is not necessarily a source of loss when iron-coke is being made.

Heat losses in the production of iron-coke are compounded of the fuel consumption in the firing chamber, and in the losses involved in quenching and cooling. In four sets of tests this amounted to between 320 and 410 kilocalories per ton of moist ore. In this process the coking action is combined with that of sintering. The thermal waste is therefore in the region of but a little lower than that found in the case of ordinary sintering under suction. It is not therefore out of the way to ascribe these losses to the sintering reaction. The reduction process therefore may be regarded as proceeding without any thermal consumption.

The economy of the iron-coke process is thus assured by the increased heat utilisation, the increased gas yield, the saving in sinter costs and the noticeable degree of pre-reduction of the charge which entails a saving of coke in the smelting process.

A corresponding calculation of the economy of the sintering process in the iron-coke production method has been published by H. Barking and C. Eymann in the periodical "Stahl und Eisen". The sintering costs are lower than that in the normal suction sintering process. Finally it is of importance to note that iron-coke can be prepared from a pure gas-coal and this increases the range of fuels which can be utilised for the smelting of iron.

By: Prof. Dr. E. Hofmann of Berlin-Charlottenburg

Since the turn of the century the pig iron production of the world has increased from an annual 40 million tons to more than the quadruple amount. The previous double production up to about 80 million tons until 1913 before the first Great War, consisting of the production of the most industrial countries of that time like USA, Germany, England, France, Belgium and Luxembourg, is mainly due to the improved gas cleaning plants as well as to the improved heating technics and operating economic conditions (Figure 1). In consequence hereof it was possible to increase the furnace measures as well as the number of furnaces continuously in the above mentioned countries. The further increase up to 167.5 million tons per year during the second period until 1953 is a consequence of the metallurgical progressive development of the blast furnace process and a more economic utilization by a suitable preparation of the ore. In addition thereto, there were the efforts of many other countries like Russia, Japan, Canada, India, South Africa and Australia for an independent industry.

With reference hereto, a further increase of the production for meeting the still high demand of the world for steel can only be obtained - as there is a high raw material consumption - by using more and more fine ore as an additional raw material for the smelting processes. With regard to the fact, that in many countries there are no deposits of cokeable coal at all, whilst in other countries the deposits of cokeable coal - which are of high importance for the production of suitable, resistable blast furnace coke until now - become more and more less, another process should be started which renders possible the production of a technically valuable pig iron in an economic way, for countries having non-cokeable coal at their disposal. Although the fuel consumption for the agglomeration of fine ore can be delivered by the sintering and pelletizing plants - being developed to a considerable throughput capacity during the last both decades - with waste coke and fuel of low value, in all cases the fuel consumption of the blast furnaces themselves is to be covered by a good blast furnace coke. Even the use of electro low shaft furnaces with electric power for the production of pig iron in countries having cheap current at their disposal will not be a real solution. As to R Durrer the erection of the above mentioned electro low shaft furnaces will be favorable at those places only, where the costs for the production per 6 KWH are not higher than for 1 Kg coal.

Already more than 10 years ago O Diettrich of Klockner-Humboldt-Deutz AG, Koln-Deutz, has recommended the single stage low temperature carbonization of so-called combined ore-coal briquettes in a short, rectangular low shaft furnace of about 3 - 5m shaft height only. After a test series carried through in a successful progression since 1952 in co-operation with the Demag AG, Duisburg, by the Demag-Humboldt-Niederschachtofen GmbH, Duisburg, in the meantime the so-called D H N process has been tested so far in some long time tests in a stepwise completed, half-technical pilot plant in Koln-Kalk with a daily pig iron capacity of 12 - 15 tons, that the erection of big plants up to about 200 tons daily capacity does not mean a risk any longer.

Special care has to be taken for the grain structure of the single raw material components, as the Demag-Humboldt low shaft furnace process is based on the charging of those raw briquettes only, containing all amounts of ore, fluxes and fuel, necessary for smelting and ready for charging, and tuned to a certain slag figure, depending on the sort of pig iron to be produced.

On one hand, the reducibility depending on the just prevailing stage of iron oxidation and on the other hand the slag ratio are important and decisive for the ores. Consequently, difficulty reducible ores have to be binded in a fore fine-grained form than slightly reducibles ores. Moreover the grain-sizes as well as the sorts and amounts of the various fluxes like limestone dolomite lime-hydrate, bauxite or such like necessary for a suitable slag production, have carefully to be examined with regard to their binding strength in the briquettes in accordance with the prevailing slag ratio of the ores.

Special care has to be taken for the coal, as after the carbonization the coal will have to act as a binder in order to keep together all further raw material components in the combined ore-coal briquettes.

During extensive investigations lasting for many years, the conditions were tested, on which this result could surely be obtained with the various sorts of coal.

With reference to the favorable results of the in-tamping of coal mixtures of smaller baking capacity in coke ovens for the production of well-carrying blast furnace coke, there was the change-over to the briquetting of more slightly baking gas and open burning coal in order to obtain a briquette - by the use of high super-charging pressures at the presses - resulting sufficiently strong carbonization coke after carbonization in the Fischer-retort.

During these tests it was clearly found out, that in addition to the super-charging first of all the period of carbonization has a decisive influence on the resistibility conditions of the produced carbonization coke. During short periods of smelting of less than 30 minutes there were obtained not only the required values of resistibility but also an increased tar output.

In case of highly sweeling and baking coal, which show the unfavorable condition of sticking together after the carbonization and disturbing the equal flowing of gas through the furnace square section during the throughput of the briquettes in the furnace, the baking capacity can be reduced by the addition of blast top furnace dust, very fine ore and coking duff resp. non bituminous coal. The addition of low shaft top furnace gas will be most economical, as this contains carbonization coke already beside of fine ore components; by its equal distribution in the briquet this carbonization coke will be the support of the structure and will have a good influence on the coming carbonization in the furnace.

Consequently it is possible to use all normal pit coal - from the slightly baking coal up to the highly baking coal - by a short period of carbonization for the DHN process.

Now it is quite obvious, that just with regard to the necessary short period of carbonization of about 25--30 minutes only, the flushing gas process will be the only suitable one of all well-known carbonization processes, and it is carried through most easily and effectively in the low shaft furnace itself where high amounts of gas with high contents of perceptible heat of the respective temperatures are at disposal, -- and -- as has been proved by the smelting tests -- the carbonization period will last for about 25--30 minutes only in the low shaft furnace. Such a high carbonization speed could only be reached in a special carbonization furnace with extraordinary high costs for equipment.

Finally, the binder is of certain importance; in first line it is used as pitch, tar-pitch-smelting, bitumen, sulphite liquor and suchlike, for securing the necessary storing-stability and resistibility of briquettes -- as to our experiences of about 30--40 kg resistibility in raw briquettes -- until the real carbonization process becomes effective in the furnace. Moreover, the cold raw briquettes, charged into the furnace at about 360--400° C top furnace gas temperature in order to avoid the discharge of carbon, have to stand this temperature interval without bursting, crushing or sticking together.

Only in this way there will be the guarantee, that after the throughput in the furnace the briquettes will arrive before the tuyeres in a good condition of shape, porosity and reactivity, in order to reach the full efficiency of their well-increased surface -- won by the intimate mixture of all fine-grained raw material components -- with the still existing porosity and the highly accelerated metallurgical reactions.

As the former tuning of raw briquettes to certain values of resistibility does not give sufficient information on their real behaviour in the furnace, they are successfully being tested in an briquettes checking apparatus, especially developed by the DHN, in which the briquettes -- produced as to various views and with various binders -- will be exposed as far as possible to the real reactions occurring during the throughput in the upper carbonization section of the low shaft furnace.

This is a checking apparatus, which mainly consists of a well-insulated, small electrically heated furnace, through the inside of which top furnace gas can be transferred if wanted.

By means of a movable weight on a weighing bar with a heatresisting pressure stamp any pressure on the briquettes can be tested and the behaviour of the samples with regard to compression can be obtained exactly from a notofying writer.

As to our experiences, at an original height of 32 mm of the raw briquettes the compression of 4 mm is possible under a loading of 4 kg without causing any disturbances in the operation of the furnace; the more, as after completion of carbonization in the furnace, in a depth of 0,8 m below the charging surface the resistibility has reached more than the double value -- compared with the raw briquette -- whilst the maximum loading in the furnace is less than 2 kg. In correspondence with the conditions of a big plant the cold raw briquette is charged at 400° C and heated up to 800° C within about 25 minutes.

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a specially developed tar-pitch-smelting of the Chemical Plant "Pluto" of the Rheinische Bergbau A.G. in Gelsenkirchen, as well as a sulphite liquor. The results of these checks under 4 kg loading are stated on fig. 2. The different behaviour of the various binders is clearly recognizable.

The most favourable values - for the raw material used in this case - of about 2-1/2 mm compression at 4 kg loading and about 30 - 35 kg cold compression strength of the raw briquettes were reached by the use 6% Pluto-tar-pitch-smelting and a mixture moisture of about 6%. After completion of carbonization the values of resistibility of these briquettes increased up to 65 - 80 kg. Despite the fact, that the sulphite-liquor briquettes did prove a good behaviour even under a loading of 4 kg and showed a compression of about 2,9 mm only, their use should be restricted to those cases, where the tar-pitch-production from the used coal is insufficient (for example anthrazite), the more, as under normal conditions those briquettes are not water-proof and consequently they cannot be stored in open weather.

Fig. 3 shows the schematic arrangement of the pilot low shaft furnace plant in Koln-Kalk. The left upper part represents the furnace and the charging arrangement. The furnace has a slight oval square section of 1,04 m² and an effective hearth area of 0,82 m² between the 4 tuyeres. The furnace has an effective charging height of 3,2 m from the tuyeres level to the top stock line level, although during the continuous operation the furnace had been operated on the average of about 2,4 to 2,8 m effective height only, and the top furnace gas temperature had been kept constantly on about 380 - 400°C for reason of a sure avoiding of C-discharging out of the tar. The charging of raw briquettes is carried through via the conveyor belts, the weighing machine, trap bucket and slide, whilst for building reasons a filling bucket with valve, Parry cone and cover serves as top closing. The gas cleaning plant consists mainly of a dust catcher, for a most possible separation of the dry dust. Then there is a pre-cooler, driven by direct current, with warm water for gas cooling to about 90 - 100° C and a mud basin thereunder, a desintegrator for a pressure increase of about 350 mm water column with a tar-discharge and -transfer, a drop catcher with Raschig-rings and a tube cooler for oil catching.

The cyclone group intended for the dry cleaning had proved to be superfluous during the operation, consequently the saved pressures were for the benefit of the whole cleaning system.

During the whole test period part of the low shaft furnace gas, cleaned in this plant to less than 0,06 g/Nm³, with a calorific value on the average of 1400 - 1450 kcal/Nm³, heated the recuperator for the pre-heating of the blast to the average of 450° C at the tuyeres, whilst the greater part was burned at the surplus gas burner and discharged into the open weather without being utilized.

Finally, the lower part of this fig. shows the blower, the recuperator, the slag utilization, the cast pig bed as well as the weighing of pig iron and dust resp. their transport.

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In order to show the ways of operation of the pilot furnace, we took as a sample of the continuous 3-weeks-test-series that part of the test, comprising 100 work hours, carried through for the production of Bessemer pig iron with 1,5% Si and 0,05% S.

As can be seen from fig. 4 the uniformity of the operation of the furnace and the products won during the last 3 days will catch the eyes at once. Although during the first 2 days after the furnace was changed on another burden, some of the S-values increased to more than 0,1%, and in one single case even to 0,198%, there was reached yet a total average of 0,057%, and during the last 3 days of even less than 0,04%. Beside of the other small alterations of the Mn-, Si- and P-contents, the relatively high C-contents of the pig iron, amounting to 4,3 to 4,5% C, are very remarkable, although they have caused very small N₂-contents of 0,003 to 0,004%.

Furthermore, the slag composition was a rather uniform one, consequently the slag rations of

$$P_1 = \frac{(CaO)}{(SiO_2)} = 1,30 - 1,32$$

$$\text{or } P_2 = \frac{(CaO) + (MgO)}{(SiO_2) + (Al_2O_3)} = 1,14 - 1,17$$

showed no important alterations.

The net calorific value of the top furnace gas amounted to an average of 1425 kcal/Nm³ at an average gas constitution of 30,6 vol. % CO; 3,4 vol. % CH₄; 0,1 vol. % C_mH_n; 3,4 vol. % CO₂; 7,6 vol. % H₂ and 54,9 vol. % N₂. Single higher variations from the average value were caused by the fact, that the terms of regular taking of samples, that means about 5 minutes after charging, for controlling the influence of progression of carbonization, had been postponed in case of these samples.

The Fe-contents of the dust amounted to 14 - 20%, and the fix. C-contents amounted to 48 - 56%.

Special care had been taken for a systematic digging-out of the furnace, cold-blown with nitrogen after the last cast in order to render possible the conclusions concerning the sectional loadings on the single layers as well as to the progression of the metallurgical processes with increasing depth.

The digging-out has been carried through in 16 different layers of about 200 mm height each, and photos were taken of the respective surfaces of the furnace square section. Furthermore, the whole sample material taken from each layer had been subdivided in 17 lots as to wall-, middle- and centre zone and was examined. This further examination consisted of the completion of petrographic briquette-cuts as well as of a chemical examination with regard to oxygen separation in the different zones and layers. Unfortunately, the various, explaining photos about the petrographic and photographic considerations cannot be shown in this report. On the other hand fig. 5 shows the progressive reduction of the iron-oxygen-compositions in the briquettes during their throughput in the furnace.

Just the evaluations of the digging-out work gave an essential contribution by their various considerations and almost agreeing results to the scientific investigation of the metallurgical processes in the low shaft furnace.

In consideration of the burden basis and raw material analyses of the charged material as well as of the analyses and amounts of the output material, detailed balances for iron, manganese, sulphur, carbon, blast, gas, dust and tar were made, the results of which are summarized in the total balance of material as to fig. 6. The fuel consumption on the average of 2191 kg coal/t pig iron, obtained from the balances, can be considered as a favourable one, since on one hand it includes the amount of fuel required in the conventional blast furnace process for sintering and coking, and, moreover, a further reduction of fuel consumption can be expected for a big plant as there is the possibility to use a higher blast temperature at a simultaneous reduction of the losses through the wall. Consequently, the fuel consumption for the only pig iron production in a big low shaft furnace - with the raw material at disposal and the pig iron required - will amount to about 1000 kg/t pig iron, with relation to coke of 85% C.

As finally has been proved by the agreement of the balances with the equilibrium examinations, based on the meditations of Th. Kootz and W. Oelsen, the D H N - process seems to be of good influence on the conditions of equilibrium, with a good manganese reduction and favourable desulphurization, at a some higher degree of saturation of pig iron for carbon as a consequence of the porosity, caused by the carbonization, as well as of the almost general equal distribution of the fine grains of ore, fluxes and fuel with their bigger surface.

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By: K Meyer of Frankfurt am Main

Sintering and pelletizing, being processes for the lumpification of fine-grained iron ores, differ as to the grain sizes of raw materials used, the kind of fuel required, and the appearance of finished products.

(I) REASONS FOR, AND IMPORTANCE OF, PELLETIZING

Since the beginning of this century, fine-grained iron ores have been given a lumpy shape suitable for blast furnace treatment. There is only a decade between the beginnings of sintering and pelletizing as lumpifying methods.

While sintering took but a few years to attain vital importance, approximately three decades elapsed before Swedish and American engineers resumed the works of Andersen (1912) and Brackelsberg (1913) and developed pelletizing for commercial use. This difference in time is chiefly due to the materials being far better suited for sintering than for pelletizing. Along with the ever increasing quantities of very fine-grained ores appearing in the market, the prospects for pelletizing have been improving, the more so as the greatest part of these ores contain a high percentage of iron and their simultaneous treatment in a sinter plant greatly lessens its capacity.

Pelletizing will develop most favorably in countries where large quantities of fine-grained ores are available, principally Sweden and the USA. For practically all the other industrial countries, the importance of sintering remains undiminished. This is especially true for Germany, as in this country the blast furnaces are fed with ores of the most different origin.

(II) RAW MATERIALS AND SITES SUITABLE FOR PELLETIZING AND SINTERING PLANTS

The fields of application for sintering and pelletizing, respectively, are limited by the absolutely different fine-ore screen analyses required for each of the two processes. This is the reason why the two methods are by no means competitive but supplementary.

While fine ores up to 15 mm are suitable for sintering, the maximum grain size for pelletizing is 0,2 mm. TABLE 1 shows typical screen analyses of fine ores. The limits between the two grain sizes are very sharply drawn. If it were intended to prepare ores of coarser grain size for pelletizing, it would become necessary to give them first the desired degree of fineness. This, however, would incur considerable costs for grinding. On the other hand, the advantages that may be expected when smelting pellets in blast furnaces, are not yet great enough to justify such additional costs. It is not usual for smelters to grind raw materials (except ground basic slag and cement). While, in general coarser fine ores will accumulate at screening and crushing plants in the vicinity of blast furnaces, fine-grained ores will mostly be found in dressing plants. This will result in a fixation of the sites for sintering and pelletizing plants, respectively: the former at the smelter, the latter in the immediate vicinity of dressing plants. These locations are desirable for other reasons as well. For instance, coke breeze and blast furnace gas, the fuels required for sintering plants, will best be obtained at smelters.

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The low mechanical strength of sinter calls for short distances between the places of production and consumption.

Matters are different for the location of pelletizing plants. The required fuel, chiefly oil, is independent from the smelter. Transport of dry, dusty material means a nuisance alongside the traveling route and entails dust losses. Wetted material, however is sensitive to frost, particularly when the respective dressing plant is located in the extreme North. Hard-burnt pellets have a high mechanical strength, and, besides, are easily transportable, from which it results that their production absolutely belongs to dressing plants. H B Wendeborn has especially outlined these special conditions.

III PELLETIZING PROCESS

On having the sufficiently fine-grained stage, two further steps are necessary before a pellet may become a blast furnace charge. The next step is to give the green pellets the required diameter. They are to be thermically hardened. For the production of wet balls (green pellets), several factors are of importance, as there are surface tension of the wetting agent, wettability of the ore, its swelling property, grain size, and grain surface, as pointed out by M Tigerschiold, H F Reich and the author. Therefore, micaceous ores and ores that had been annealed to about 1000°C once before (pyrite cinders) will not easily form balls, whereas raw materials of plastic properties will. The existence of the green pellets is only an intermediate stage in the process of manufacturing finished pellets. For further treatment in shaft furnaces or on sinter strands they need a certain minimum of strength, which will be 2 kg ($\frac{1}{3}$) chrushing strength for shaft furnaces and 1 kg for sinter strands. The higher strength can only be attained by additives or by a more thorough fine-grinding, which means higher operating costs.

For the production of green pellets, two basic ways are possible, viz.

- (a) Forming the balls from smallest particles on the snow-balling principle in drums or on discs;
- (b) Rough pressing of cylinders and rerolling them in drums

For the snow-balling principle, cylindrical drums are mostly used in Sweden and the USA; on their dimensions and effectiveness M Tigerschiold has already reported in detail. According to this report, the length of the drum is to be 2 to 3 times its diameter, its slop about 2 to 5%. Production will fluctuate, depending on the size of the pellets to be produced. Variation of rotation speed will change the diameter of the pellets. To ensure good functioning it is important that the inside of the drum shell has a certain gripping capacity, which will best be produced by a layer of the wet ore itself. Excessive growth of this layer will be prevented by a scraper. An advantage of the drum lies in its relatively high production capacity, whereas it is a drawback that the pellets, that are at the same time leaving the drum, are however, not of a uniform diameter. For this reason, it is imperative to have a screen follow the drum, by means of which the required grain size will be separated for further treatment, whilst the undersize will be returned to the drum, sometimes up to 400% of the quantity produced.

It has first of all been in Germany where the granulating disc has chiefly been used for the production of green pellets. Compared to the drum, it shows some substantial advantages, such as lower weight as well as greater variation with respect to rotation speed, slope and height of disc wall.

It is the disc that will assort the balls, discharging them over the disc wall within narrow grain-size limits. This is a function of the motion of the material. FIGURES 1 AND 2 show the path of motion in a drum and in a disc, respectively. FIGURE 3 represents the front view of a granulating disc, namely a special variant. A channel is fastened to the outside border of the disc. Balls coming from the disc will first roll into this channel, where they can be covered with another material, which is another essential advantage, as compared to the drum. The sizes of the green pellets vary according to slope and rotation speed. FIGURE 4 shows pellets produced on discs.

For iron ores the productive capacity will fluctuate between 3 and 25 tons per sq. m. per 24 hours. Discs of 2, 3, 4, 4.5, 5 and 5.5 m diameter are built. Capacity, however, does not increase proportionally to the enlarged surface of the disc. The productive capacity per square metre of a 5-metre disc will only reach 70% of that of a 2-metre disc. This results in setting a limit to the enlargement of the surface. FIGURE 5 is a section of a 5-metre disc in operation for the pelletizing of flotation cinder; FIGURE 6 shows the removal of green pellets. This disc was developed by Lurgi, in collaboration with Maschinenfabrik B Beumer. Several discs are now in operation. Experiments for an improvement of efficiency of the disc surface were various, intensive development is still going on. It seems untimely however, to publish promising partial results at this early date. Up to the present, practically all of the fine ores suitable for pelletizing could be formed on discs.

When looking for machines offering both a high throughput capacity and a fairly uniform size of pellets, the vacuum press (known from ceramic industry) was also minutely tested for a longer period. Reference is made to the article by A Stirling and to that by G Sengfelder. The results of these tests are approximately as follows: only fine ores of sufficiently plastic properties can be formed on presses; the porosity required for reaching a high degree of oxidation during the hardening process is lower than is the case with balls formed on discs; this results in an increase in bivalent Fe content and in a decrease in strength; the vacuum press is much more sensitive towards fluctuations in moisture content than are the drum and the disc; moreover, it is extremely sensitive to hard foreign particles.

The last step in the production of finished pellets is the hardening process applied to attain the necessary mechanical strength. This is done by cautious heating up to the point where sintering is beginning. Details on hardening possibilities have been given by M Tigerschiold.

Until lately, pellets were exclusively produced in shaft furnaces which are round for smaller capacities and rectangular for larger ones. The dimensions commonly used in the USA are 7 ft. 14 ft. 34 ft. in Sweden 6 ft. 10 ft. section.

The green pellets are charged into the top and discharged at the bottom. Heat supply is done by strongly heated combustion gases. Up to now, the capacities reached were about 350 to 400 tons per shaft furnace that is often composed of several of the above units. Recently, however, mention was made of remarkable increase in capacities reached by an increase in blowing rates, stepping up to 800 - 900 tons in American furnaces. More details on shaft furnaces have been given by M Tigerschiold.

Shaft furnaces have proved advantageous for pelletizing magnetites, because a considerable part of the required heat is supplied by oxidation to hematite. But shaft furnaces are badly or not at all, suited for hematites.

or such ores that have a small range of temperature between their sintering and melting points. There are other disadvantages, too, such as the difficulty of maintaining a uniform gas passage within the furnace, and the danger of disintegration or agglomeration of the charge. It will, therefore, be understood that tests have also been made for the purpose of using the sinter strand for the hardening of pellets. Developing tests were simultaneously being made in the USA and in Germany. Up to now, one largescale plant for a capacity of 1000 tons per 24 hours was built by the firm of McKee at Babbit. Lurgi has been carrying through such tests since 1949 and is now erecting a 1100-ton plant. All results were at first established in a laboratory plant and then controlled in a pilot plant.

All these tests have been guided by the desire to pelletize practically all materials of finest grain size on the sinter machine, making it a principle to exclude all additives of salts or bentonite, as are needed for shaft furnaces. Neither has the material been mixed with solid fuel. The test results gained up to now clearly show that it is possible to produce pellets up to 30 mm (1-1/4").

Special arrangements in the design of the burning machine as well as the feeding and at the operational control have prevented direct contact of the hot gases with the side walls and grate bars of the pellets. The use of suitable alloys means an additional safety factor for trouble-free operation. The application of the sinter machine as a pelletizing apparatus is based on the devise: safety of the machine first. Shockless feeding of the pellets onto the machine is very important, for which purpose special devices have been created. Drying and burning of the pellets is done by sucking; cooling is done by pressing.

FIGURE 7 is a scheme of a Lurgi pelletizing plant.

As compared to the shaft furnace, the sinter machine has several remarkable advantages: It can be built in one unit for capacities up to 1500 tons; temperature control is easy, its adjustment precise and adaptable to the properties of the respective ores; the low depth of layer and relative repose of the pellets do not require such high initial strengths; badly burnt material can easily be screened off and recirculated to the process; heat losses can be reduced to a minimum by circulating part of the gases; the range of application is practically unlimited; much less is required of the range of the grain sizes than is the case with shaft furnaces.

(IV) A PELLETIZING PROCESS FOR PYRITE CINDERS, SIMULTANEOUSLY CLEANING THEM OF COPPER, LEAD, AND ZINC. - VUOKSENNISCA-IMATRA PROCESS

There is a possibility of carrying through chemical reactions simultaneously with a thermal treatment of green pellets. This proves important wherever the value of high-priced raw materials is lessened by impurities. This is especially true for pyrite cinders poor in phosphorus, but contaminated by Cu, Pb, and Zn. By the classical process, cleaning the materials of nonferrous metals is possible where contents are high, but uneconomical where they are low. In its Works of Imatra, the firm of Vuokesenniska AB developed a thermal process for the utilization of such cinders. This process was taken over by Lurgi for sale.

FIGURE 8 is a scheme of this process. It has been made possible to pelletize and clean flotation pyrite cinders by one step.

TABLE 2 gives some values before and after pelletizing, respectively. Intensive developmental work has helped to reduce operational costs to such a degree that this process appears to be absolutely economical. The plant at Imatra now treats about 130 - 140 tons of flotation pyrite cinders per day. It was taken in operation in 1952 already.

(V) APPLICATION OF KNOWLEDGE GAINED AT PELLETIZING TO GENERAL DOWN-DRAUGHT SINTERING

The question that arises is: do sintering and pelletizing contrast or is there a chance of usefully applying any of the knowledge gained at the development of one process to the other process? The latter question should be answered in the affirmative, as doubtlessly, there are, apart from the optimum grain distributions for sintering or pelletizing, respectively, many other grain distributions that will suit neither group. This will principally be the case whenever different kinds of ore, whose properties are varying, have to be treated.

With various ores of a high percentage of fines it has proved possible substantially to increase the efficiency of sinter grates, when the thoroughly mixed mass of components was, for a longer time, subjected to rerolling in a drum. But it is also possible to separate finest ores from coarser lots, for instance by screening or air sifting, and to sinter the coarser section effectively, and to pelletize the fines fraction.

At the sintering process the required fuel, that is of a solid form, is usually mixed with the ore and ignited. At the pelletizing process the heat supply is done, by the greatest part and sometimes altogether, by means of heated gases from the outside no matter whether it is question of shaft furnaces or sinter strands.

Since 1949/50 Lurgi has experimenting on combining both burning techniques and FIGURE 9 is the respective scheme. An extended ignition furnace covers part of the sinter grate, so that the hot gases are sucked through the sinter mixture for a longer time than is necessary for ignition alone. The result obtained is surprising in many respects:

1. Depending on the effect exercised by the hot gases, the total fuel requirement will decrease to 75 to 80% of the requirement of fuel when using coke breeze alone.
2. The degree of oxidation of sinter increases from 90 to 92% to 95 to 98% and approaches that of the black burnt sinter.
3. The capacity of the sintering machine decreases when prolonging the effect of the hot gases.
4. The sinter is not more slagged to such a high degree and yet the quantity of return fines is smaller.
5. Applying this process means a better adaptation of the sintering to the respective fuel situation. The first sinter plant of this kind was built by Lurgi in co-operation with the firm of Arbed at Saarbrücken-Burbach. The sinter grate of an effective suction area of 50 sq.m. and a daily capacity of 1000 tons has been in operation since August 1952. In 1954 a further machine was switched over to mixed firing at the same Works. Since May 1954 a

sinter grate working by this process is also in operation at Huttenwerk Phoenix, Duisburg. Several other Works in France an Italy have decided on changing over to mixed firing.

By Dr.-Ing. D. Fastje of Watenstedt-Salzgitter

The metallurgical principles of the Krupp-Renn process and its operation have already been reported in "Stahl und Eisen" 69 (1949) pp 319 to 325 so that only the more important features of this process need be recapitulated before the new developments are considered.

The Renn process is a reduction process for iron ore which uses a revolving kiln furnace, and in which the iron oxides are first reduced by means of the fuel mixed in with the charge to an iron sponge at a temperature of between 600°C and 1000°C. This sponge is then converted in a higher temperature zone to larger bodies of low-carbon iron which are known as "Nodules" (Luppen), caused by the sponge being welded. At the same time the gangue minerals in the ore form a semi-liquid slag in which the nodules remain embedded. The separation of metal and slag is completed by mechanical methods operating in conjunction with the kiln. The final products are gangue-free nodules which contain between 93 and 95% Fe and which will contain varying quantities of sulphur and phosphorus according to the analyses of the ore and fuel being utilised, but which on the other hand contain no manganese or silicon as these elements are not reduced from their oxides at the relatively low operating temperature of the Krupp-Renn process. For the same reason the carbon content of the nodular iron produced is also relatively low.

Fig. 1 shows schematically the method of operation of the process. The ore which according to its reducibility must be crushed to a particle size of 2 to 20 mm diameter, is mixed with finely crushed fuel and fed to the inlet end of the kiln where it passes along in the opposite direction to the combustion gases which are generated by a flame located at the outlet end of the kiln. In the first zone of the kiln, which is known as the pre-heat zone, a gradual warming-up of the charge takes place and moisture and water of crystallisation are driven off. In the second zone of the furnace, where the temperature is between 600°C and 1000°C the iron oxides are reduced to iron sponge by the reducing action of the carbon present in the fuel. The carbon monoxide generated inside the charge by this reaction burns in the free space above and thus supplies enough heat for reduction to proceed continuously. In the final zone of the furnace, in which the temperature is raised to about 1250 to 1300°C by the help of a pulverised coal- or ore flame, there occurs a welding of the finely divided iron sponge into nodules due to the combined effects of the oxidising combustion gases and the turning action of the kiln. A heavy and viscous slag is formed from the gangue constituents of the ore, which slag has only a low iron content, and the nodules are carried to the outlet of the kiln in this slag. The material issuing from the outlet of the kiln is quenched with water and cooled and is then passed through suitable crushing machinery in which the nodules remain unaffected, and they are then removed from the finely-ground slag by means of screens and magnetic separators.

One advantage of the Krupp-Renn process is that it is largely independent of the quality of the fuel used as a reducing agent, that is to say practically all the so-called "waste" fuels can be used for this purpose such as coke breeze, brown coal semi-coke (Braunkohlenschwelkoks), fine anthracite, low-grade coal slack etc. — The ash content of the fuel

used plays no effective part in the reaction, only the volatile content of the fuel must be low as these gases cannot be usefully employed inside a Renn kiln. The sulphur content of the fuel is of considerable importance as a part of this sulphur is transferred to the nodules and the low operating temperature and the very low basicity of the slag offers no hindrance to sulphur pick-up by the iron.

The most important metallurgical features of the Renn process are as follows:

1. The process operates almost exclusively with slags which, in the range of temperature at which iron exists as a tacky, low-carbon mass, i.e. iron can form nodules; is sufficiently heavy and viscous as to be able to hold these nodules in suspension. As a general rule only slags with a silica content of 55 to 70% possess the necessary degree of viscosity.
2. The slag volume should not fall below 800 to 1000 kg per ton of nodules, because otherwise the iron will coagulate into balls too great a degree and this will cause difficulties in the operation of the kiln.

Thus the most important application of the Renn process is already indicated: It is particularly suited for the treatment of low-grade highly siliceous ores which are uneconomic to smelt directly by present day techniques in the blast furnace and which are unsuited for beneficiation or any of the usual ore-preparation processes prior to blast furnace treatment.

From this there arises the question whether the Renn process should be considered only as a method of pre-treatment of low grade ores, i.e. as a first stage in the production of iron in the blast furnace or whether it can be considered as being already a member of the real iron-producing processes. This question must be answered in one way or the other according to circumstances. Nodules with a high phosphorus and sulphur content, which are produced when a phosphoric ore is reduced with a sulphur-containing fuel, must be first charged to the blast furnace if an iron is to be produced which can be converted to steel by the normal known methods. This is economical particularly in cases where a blast furnace plant is already in existence. In effect this raises the capacity of the blast furnace since the production of pig iron from Renn nodules entails no reduction or slag formation and only 200 kg of blast furnace coke are required per tonne of pig iron produced.

In other cases, where the quality of the ore and the low sulphur content of the fuel are such that the production of nodules low in phosphorus and sulphur is feasible, then these nodules can be processed directly into steel in the electric arc furnace or in the open hearth furnace, without the cost increase involved in first producing a pig iron. In this instance the Renn process is not a pre-treatment method, but is an iron-producing process, in fact a method of generating a form of "artificial" scrap. Unlike the ordinary grades of commercial scrap this Renn nodule "scrap" possesses the advantages of easy chargeability and freedom from the unwanted "tramp" elements such as Copper, Nickel, Chromium, Molybdenum.

At the present time developments are taking place which will tend in the future to make the Renn process more important as an iron-producing method. The technical development of the arc furnace and its increasing size, together with the increasing efficiency in the generation of electric power make it likely that the large electric arc furnace will be found to be the most suitable metallurgical apparatus for the direct conversion of Renn nodules into steel. The advantages of supremely easy charging and melting which are obtained when Renn nodules are used may well offset the fact that owing to the sulphur contained in this material a higher slag volume than normal must be carried. With the use of the oxygen lance the electric arc furnace is becoming more able to undertake refining operations, and thus it is not impossible that ways may also be found to process high-phosphorus Renn nodules direct to high quality steel in the arc furnace.

These technical developments, which are already well under way, in any case mean that it is advisable when considering the construction of new iron and steel plants to take into consideration more and more the possibilities of the Renn kiln — large electric furnace combination which both technically and economically in the saving of first cost possesses some definite advantages when compared with the orthodox blast furnace and open hearth resp. basic converter system.

The industrial development of the Renn process began rather more than 20 years ago with the erection of a large experimental installation at Essen, Germany, in which up to the outbreak of the second World War low grade German ores were treated. In the former Japanese areas of Manchuria and Korea there were 4 Renn kilns installed by the end of the 1930's and 2 such kilns were in operation on the Japanese mainland. During the war all these plants were developed and extended and new plants were put down in other localities so that according to information received there were 24 Renn kilns operating under Japanese supervision by the time the war ended. During the war also one Renn furnace came into operation in Czechoslovakia and in Germany the big Renn plant at Watenstedt was erected.

Since the end of the war the Czech plant has been expanded to two kilns and a Renn plant installed in North Spain which will come into operation in a few months. A plant comprising two kilns has been constructed and put into operation at Unterwellenborn in the East Zone of Germany.

The plant at Borbeck, and the largest and most important Renn installation in Germany at Salzgitter-Watenstedt both fell victims to the post-war dismantling programmes. A few weeks ago, however, the decision was taken to re-construct the Watenstedt plant in a more up-to-date form and with increased capacity. A comparison between the arrangement of the kilns before dismantling and the newly planned system is shown in Fig. 2.

The original plant before dismantling consisted of 3 furnaces of 4.2 metres diameter and 70 metres length with an annual capacity of about 450,000 tons of ore. At that time a raw ore containing between 26 to 28% Fe was treated, the annual production of nodules being between 110,000 and 120,000 tons. The kiln charge consisted of ore, fuel, an intermediate product arising from the slag milling plant and flue dust, and this charge was extracted from bunkers by plate conveyors and weighed out in scale lorries. The constituents of the charge were mixed in revolving feeders and the prepared charge then fed into the inlet end of the kiln by means of a bucket elevator. This intermittent charging system is to be abandoned in the new project,

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primarily in order to dispense with the arrangement of weighers, feeders and bucket elevators which require a great deal of maintenance. In the new installation the preparation of the charge will be undertaken in a manner similar to that employed in sinter plants, rotary table feeders and automatic weigh-feed belts being employed, while the prepared charge will be taken to the mouth of the kiln by means of belt conveyors. The kilns themselves will be enlarged, 2 kilns of 4.2 metres diameter and 95 metres length and one kiln 4.6 metres diameter and 110 metres length being installed. These three new furnaces will together be able to treat some 600,000 tons of ore per annum.

It is to be noted that particular consideration has been given to the question of the type of ore which will be processed in the new Renn plant. A wet concentration plant has been erected which can produce a concentrate from the ores of the Salzgitter area. The local lean, siliceous ores contain between 25 to 28% Fe and 22 to 27% SiO_2 , and as a result of technical improvements in the concentration process and the introduction of new methods the concentrate contains 39% Fe and only 14% SiO_2 . In practice however this concentrate has to be diluted with "middlings" containing only 30 to 38% Fe, but it is possible that in the future these "middlings" may be treated directly in the Renn plant. This will have a double advantage, on the one hand owing to the increased iron content of the charge the net output of Renn nodules will be increased to about 180,000 to 200,000 tons p.a., which will raise the economic efficiency of this process, while on the other hand the wet concentrators attached to sinter plants and blast furnaces in this area will be able to produce a concentrate containing more iron, i.e. with a Fe content of 40.5 to 41%, with a correspondingly lower silica content. In this manner the new Renn plant will become an important factor in the entire economy of the Salzgitter orefield. It is not unlikely for example that much of the fine ore now produced here in the ore crushing and screening plants, and which at present must be either sintered or pelletized before charging to the blast furnace may in future be processed into Renn nodules before blast furnace treatment. By the choice of this system there is also the possibility that at the same time the dust and the mud from dust-catching plants, which are not altogether very suitable for sinter plants, may be treated and their lead and zinc contents recovered from the flue gases issuing from the Renn kiln.

The Renn process should also be tested for the treatment of the American taconite ores, which in the future must be further exploited in order to make up for the depletion of reserves in the Lake Superior deposits. These are very low-grade ores with a high silica content and they can only be made suitable for blast furnace treatment after a costly operation of fine milling, often with a poor recovery. Beneficiation is only possible in a taconite in which the iron is combined as magnetite, which can therefore be enriched by magnetic separation. In the case of the non-magnetic taconites, in which the iron is held as carbonate, haematite or in a silicate complex, no suitable pretreatment process has yet been devised.

Some years ago the Renn process was tried in America on a poor ore containing 27 to 32% Fe. In the processing of a taconite, however, there is one grave disadvantage to be contended with, namely that because of the very high content of silica it is necessary to add substantial quantities of lime or alumina to the charge in order to produce a workable Renn slag. This

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entails a relatively large volume of slag, which in spite of its low iron content can account for a loss of 3 to 5% in recovery. In addition the capacity of the Renn kiln is reduced owing to the large volume of slag carried.

A higher degree of efficiency due to the lower slag bulk and greater throughput may be obtained in the case of processing Salzgitter ores, if instead of charging the raw ore the concentrates and some of the middlings from a magnetic or similar beneficiation plant be charged to the Renn kiln. It is for example thus possible that in the case of a magnetic taconite the fine grinding should not be taken so far as normal, but instead a relatively coarse-mesh product should be sufficient and therefore along with the concentrates and tailings a lower-grade "middlings" could be recovered with an iron content of perhaps 35% Fe. This intermediate product might then be processed into iron nodules in a Renn kiln. This would reduce the total crushing costs and it may also be expected to improve the overall iron recovery.

In non-magnetic taconite deposits there are bodies of ore which possess a certain magnetite content and which can be enriched by magnetic separation, but not sufficiently for blast furnace smelting. If, however, the Renn furnace is proposed instead of the blast furnace then an enrichment to 35 to 40% Fe suffices, which is possible in a number of cases.

The wholly non-magnetic taconites, which are today untreatable can probably be made suitable for Renn furnace smelting if they are mixed with concentrates or high-grade middlings from a magnetic taconite concentrator before being charged to the kiln. Provided this mixture contains about 40% Fe it should be smeltable. A thorough investigation of these possibilities is clearly called for in order to understand the fundamental technicalities.

To assess the economics of the Renn process the following data may be used as a general guide —

Under present-day German conditions the first cost of the construction of a Renn plant for the annual production of 400,000 tons of nodules from about 1.2 million tons of ore with 35% Fe content amounts to about 90 million Deutschmarks (8 million £). This plant would consist of 5 kilns of 4.6 metres diameter and 110 metres length. Allowing for depreciation and interest together at the rate of 15% this amounts to a capital charge of roughly 34 DM (£ 3) per tonne of nodules produced.

Taking the price of raw ore, "middlings" ore concentrates at 12 DM per ton (about 22/ — per ton) then the ore cost works out at about 36 DM (£ 3.3) per ton of nodules. A consumption of "waste" fuel at the rate of 1000 kg per ton of nodules (20 cwt per ton) is to be expected, at a price of about 40 DM (£ 3.7) per ton.

The operating costs, i.e. power, maintenance, wages etc. will run at the rate of about 30 DM (£ 2.75) per ton of nodules.

The total costs therefore for the production of Renn nodules from a low-grade acid ore in a large modern plant will amount to about 140 DM per ton, i.e. about £ 13 per ton.

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This cost potentialities of the Renn process under various conditions and in different places, particularly in those cases where a further processing of the nodules in the large electric arc furnace can be envisaged. If the conversion cost in such a furnace be assessed at 80 DM (F 7.3) per ton of ingots produced, then the production of electric steel at a cost of 220 DM per ton of ingots (F 21 per ton) becomes a practical possibility.

By Dr.-Ing. F. Wesemann of Dusseldorf

I should like to report on investigations on the testing of the mechanical strength of metallurgical coke conducted recently at a number of blast-furnace plants. The mechanical strength of coke is measured by the drum test developed some thirty years ago by Simmersbach. Before the last war, a German standard (DIN) specification was laid down for this test. The method of evaluation in this specification, however, underwent several modifications of the so-called Ilseder index although these have recently become controversial.

In the past, random determination of the drum index was usually regarded as sufficient, but in recent years more and more blast-furnace plants have changed over to continuous testing of the coke on delivery. Continuous testing, however, requires mechanisation of the preliminary screening. Preliminary screening of the coke with a fork with prongs 50 mm. (2 in.) apart, according to the DIN specification, as well as the screening of the coke after the test to 40 mm. (1 — 9/16 in.) was previously done by hand. W. Wolf, at the blast-furnace plant of the Westfalenhütte, recently developed a mechanical screening method which is now being employed at several blast-furnace plants. The device consists of a number of screening boxes stacked one above the other. These boxes 1 sq.m. (1.2 sq. yards) in area have plates with circular holes ranging from 100 to 10 mm. (4 inches to 3/8 in.) in diameter. The stack stands on an electrically driven jolting table which is vibrated at a frequency of 1000 cycles per min. and an amplitude of 3.67 mm. (0.144 inch). The electric drive is time-controlled and switches itself off after 40 seconds. A great number of tests showed that for a given coke the percentage of the various fractions is the same with mechanical screening as when it was done by hand. When using mechanical screening in combination with the drum test the instructions in DIN 51,712 were followed, with the sole exception that the fraction larger than 60 mm. (2 — 3/8 in.) was used instead of that obtained with the hand fork with prongs 50 mm. (2 in.) apart.

The fact that mechanical screening reduces the total time needed for the drum test and permits the screen analysis of metallurgical coke to be made without additional manual labour has led to remarkable progress in determining the indices characterizing the mechanical strength of coke (Fig. 1).

The screen analysis of coke as found by mechanical screening before drum-testing is given on the left in Fig. 1 where all grades above 60 mm. are designated "D". On the right are given the letters denoting the size ranges used to represent the strength indices of the drum-tested coke, viz.

- A = percentage above 60 mm.
- B = percentage from 40 to 60 mm.
- C = total percentage below 40 mm., and
- L = percentage below 10 mm.

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These letters are used to characterize the mechanical strength in the upper and lower part of Fig. 2. The former strength index in the upper part of the diagram is expressed by A plus B, i.e. it represents the total quantity of the sizes above 60 mm. In the next line, the original Ilseder index A minus C, i.e. sizes above 60 mm. less the amount found below 40 mm., and finally in line 3 the new Ilseder index which consists of the size A above 60 mm. after drum-testing, divided by D the percentage above 60 mm. before drum-testing; from the quotient is subtracted the amount C below 40 mm. obtained after testing. All three indices, however, do not take into account the fractions below 10 mm., which should not be omitted since this is an important factor in blast-furnace operation. Furthermore, the mathematical expression of the new Ilseder index is in many respects not satisfactory.

A suitable characteristic for the strength of coke should contain three main components, viz.

- (1) D = percentage above 60 mm. before the drum test
- (2) A = percentage above 60 mm. after the drum test, and
- (3) L = percentage below 10 mm. after the drum test.

The three components D, A, and L represent a selection and summation of the original values obtained in mechanical screening and drum-testing. It is obvious that if all the actual values obtained in the whole test, i.e. all the screenings rising by 10 mm. stages were incorporated in a formula, better information could be obtained than if only three factors are employed, but graphical or numerical evaluation of such a formula would be too elaborate for practical use. The new index, k, consists therefore of only the three components A, D, and L, in the form shown in the lower part of Fig. 2.

$$k = \frac{A \cdot D}{100 \cdot L}$$

The product (A . D)/100 representing the percentages of the grades above 60 mm. before and after drum-testing, should be large, whereas L, the grade below 10 mm. after testing, should be as low as possible. As L is in the denominator, the value of k increases with decreasing L.

(A . D)/100 is plotted against L in a coordinate system; i.e. k is represented by a point in the diagram. The value of the index k can be read directly from the diagram without having to use a slide rule. A number of tests, e.g. taken over a month, gives a "cluster" of points; two of these "clusters" for different cokes are shown in Fig. 3. It can clearly be seen from the diagram that the two clusters are well separated and experience has shown that this type of diagram discriminates sharply between the strength properties of different cokes.

Fig. 4 shows the results obtained for five types of coke, but for the sake of clarity only the cluster boundaries are plotted. It must, however, be stressed that the position of the cluster in the diagram does not in any way appraise the coke or its suitability for use in the blast furnace. The particular position of the cluster in the diagram is not governed solely by the mechanical strength of the coke but mainly by the screened fraction of the coke as-delivered. If, however, a record is kept over a long period (e.g. a month) of the test results of the coke from one

Approved For Release 2001/09/07 : CIA-RDP80-00926A007900040001-2
supplier certain conclusions regarding the average quality and uniformity of the coke can be drawn. In these long-period records, not only the position but also the area of the cluster are regarded as the criteria for quality. Evaluation of a large number of test results has shown that changes in the coking practice of one and the same supplier, such as variation in composition and grainsize of the charge, coking time or temperature, are clearly reflected by the displacement of the point in the diagram.

Further research is, however, necessary in order to establish in what way blast-furnace operation is affected by movement of the points and by changes in the area of a cluster in the diagram, or, to put it another way, in which part of the diagram must the cluster representing a certain coke lie in order to be regarded as good material for the blast furnace.

As my time is limited, further advantages offered by this method of testing cannot be discussed. I will only mention the correlation between the new coke index k and the results of the shatter test or Wolf's abrasion index; further, the influence of the sampling place and of transport and loading and unloading on the index k as well as the brittleness of the coke, which will be taken into account by modifying the index k.

The Coke Properties Sub-Committee of the Committee on Coke Production cooperates closely with the Steinkohlenbergbauverein (Coal Mining Companies Association) and we trust that this close cooperation will be beneficial to both parties in their effort to improve the quality of metallurgical coke which in turn will ease the operation of the blast furnace.

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